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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE UNITED VERDE COPPER CO., CLARKDALE, ARIZ.[1]

BY

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INTRODUCTION

This paper, describing the concentrator practice of the United Verde Copper Co., is one of a series being prepared by the Bureau of Mines.

HISTORY

The United Verde concentrator was constructed in accordance with recommendations based on certain novel reasons which were quite different from those usually justifying the expenditure of such an amount.

Concentrator operations started in March, 1927, although the metallurgical staff had considered for several years the eventual addition of a concentrator department.

United Verde ore had always been considered a direct smelting ore and the cost of producing copper by such methods satisfactory. However, pyrometallurgical methods alone resulted in certain smoke nuisances due to the use of all sulphides in the ore as fuel in the smelting operations. Apparently it was logical to avoid smelting these large quantities of iron sulphide and to store them for the future in the hope that sooner or later an economical process would be developed that would yield a profit on this iron and sulphur.

The separation of iron sulphides from such ores by differential flotation methods had been considered, but was not even in a state of embryonic development; nevertheless, laboratory flotation tests were undertaken which disclosed that such a separation might be reasonably expected.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided, the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6343."

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Owing to the degree of dissemination of the minerals, which are largely microcrystalline, it was evident that extremely fine grinding was a ponderous problem that could be solved only by determination of costs upon a large scale. There was also considerable doubt as to the economy of a milling process that would involve lessened recoveries of silver and gold, as compared with direct smelting. The pitfalls were therefore so ominous that the metallurgical staff did not feel justified in recommending the construction of a large mill for differential flotation of direct smelting heavy sulphide ores. Believing, however that subsequent improvements in flotation metallurgy and in grinding practice would eventually solve the problem the staff sought justifiable reasons for recommending the construction of a concentrating plant that could be used economically for selected portions of the ore body and at the same time be utilized for working out the major problem of differential flotation of the heavy sulphide ores.

Those portions of the ore body containing large amounts of black, slaty looking rock, locally called "schist", yielded about 1,000 tons of ore per day which, from the pyrometallurgical point of view, was more refractory than the main mass of the ore body. Therefore, a laboratory flotation investigation of such schist ores was made. Although this investigation was entirely successful as to flotation, the predicted combined costs of flotation concentration plus concentrate smelting did not show enough profit, compared with the cost of the direct smelting of this schist ore, to justify the expense of a concentrator installation. However, metallurgical calculations did show that the effect of the schist ore on the cost of smelting the average run-of-mine ore, of which the schist ore constituted only about 25 to 30 per cent, was important. Metallurgical data and calculations indicated that if the schist ores were mined separately and then concentrated the smelting costs on all the other ore from the ore body would be materially reduced. About 105,000 tons of ore per month was being produced by the mine; of this, 30,000 tons could be classed as schist. Calculations showed that 15 to 25 cents per ton could be saved at the smelting plant on the total ore mined if only the 30,000 tons of ore per month were concentrated for removal of the black schist mineral. A recommendation to build a concentrator was then approved and executed; as a result, expectations have been more than realized in the handling of the schist ore and the company has been provided with excellent means of studying the major heavy sulphide problem.

THE PROBLEM

The United Verde ore bodies represent a replacement of pre-Cambrian schists by sulphides. The degree of replacement is variable, as is also the character of the replacing sulphides, a condition that necessitates classification of ore both with respect to schist replacement and the quantitative relations of mineral types in any given area. Thus, ores are classified locally as schist, siliceous, siliceous massive sulphides, and massive sulphides. The last are further classified with respect to valuable metal content as zinc massive, copper massive, and copper-zinc massive. The term massive sulphide as used herein implies that pyrite is a major constituent of the ore; it does not refer to massiveness of the valuable minerals. Such a classification is necessarily broad and concerns only outstanding mineralogical characteristics. All of the ores contain schist and quarts, with sulphides of copper, zinc, and iron in variable amounts.

Mining operations, though attempting segregation of ore types, can not be so closely conducted that mixing types can not occur, and the influence of such mixing is reflected in the character of the schist ores and massives in general, as delivered to the reduction works. Conditions within the ore body dictate the mining methods followed and bear directly upon the problem of segregation. Methods employed may be classified broadly as underground and open pit, and both are concerned with all ore types. The latter operation is of particular interest, because it involves ores that for many years were unavailable, being in a zone of early underground workings bulkheaded off on account of an uncontrollable fire. Though some ore may be found in place in this zone, much of it has been displaced by subsidence of old workings and is mixed with the stope filling of those workings. The whole is chemically altered by fire conditions, the alteration ranging, in the sulphides, from superficial tarnishing to complete calcination. The schist ore in this area is not as extensively altered as the massives, but it is difficult to mine it cleanly.

Initially, concentrator operations were concerned solely with the unburned schist ore. In recent months the sulphide content of the concentrator feed has been increased by admixture of massive sulphides, and at times pit schist ores have constituted a large percentage of the tonnage of ores treated. The following table reflects those conditions and offers data as to metallic values and mineralogical form of metals and gangue, as shown by analyses of monthly composites of concentrator feed:

Chemical analyses

Period	Per Cent								Ounces per ton	
	Total Cu	Oxide Cu	SiO_2	Al_2O_3	Fe	S	Zn	CaO	Ag	Au
Aug. 1927.	3.00	---	37.2	13.1	17.5	10.7	1.0	0.9	0.632	0.011
Sept. 1929	3.06	0.17	31.2	11.4	20.4	16.1	1.2	1.6	.738	.016

Calculated mineralogical analyses

Period	Per cent						
	$CuFeS_2$	FeS_2	ZnS[1]	Schist	Quartz	Hydrated Iron oxides	$CaCO_3$
Aug. 1927..	8.6	13.5	1.5	52.4	23.0	2.9	1.6
Sept. 1929.	8.9	23.2	1.8	42.6	18.9	1.6	2.9

[1] Zinc is calculated as sphalerite but is present as marmatite. Specific gravity of the ore varies between 3.3 and 3.5.

The variable nature of the ores treated, which is shown by the above table, prevents presentation of a composite picture of ore character. It will be noted that in the above table schist has been classified as a mineral. This is permissible under local conditions because of the chemical uniformity of this gangue material. An approximate analysis of the schist follows:

	Per cent
SiO_2	28.0
Al_2O_3	25.0
FeO	13.0
CaO	0.3
MgO	17.0
Ignition loss	10.0

In the above ores copper is present principally as chalcopyrite, although minor amounts of covellite and bornite have been noted. Oxide copper minerals are negligible in the ore derived from underground mining but reach serious proportions, from the ore-dressing standpoint, in the ores from the fire zone of the mine. Inasmuch as the oxides present undoubtedly represent the products from partial calcination of sulphides, concentration of such minerals in the finer sizes of the mill feed would be expected. The oxide content of the ores milled in September, 1929, was above the average, and the plus 100-mesh size portion of the mill feed showed 0.118 per cent oxide copper and the minus 100-mesh portion 0.31 per cent. The moisture content of the ore is low, seldom exceeding 1.5 per cent, and offers no difficulties in milling. Aside from the alteration due to fire conditions in the pit no other alteration has been noted, incident to mining methods.

The degree of dissemination of the valuable minerals in the ores of the several types is variable. In general, it may be classed as extremely fine in comparision with most copper ores treated in the Southwest. In the schist ores, dissemination of the sulphides in the gangue is such that minus 65-mesh grinding effects liberation. The association of the chalcopyrite with the marmatite and the pyrite is, however, more complex, particularly in the massives. With a need for the separation of these minerals finer grinding becomes necessary.

The extent to which such grinding must be carried to effect unlocking ranges for the different ores from reduction through a 200-mesh opening (0.074 mm.) to reduction through a 300-mesh opening (0.046 mm.), and on some ores locking is still in evidence in the 19-micron, (roughly, 800-mesh) size. The screen analyses of mill products shown in Table 1 reflect conditions of grinding and dissemination of copper values for September 1929. During that month admixture of massive sulphides with the schist ores was marked, and the percentage of pit ores large. Metallurgical results are therefore influenced both by degree of dissemination of the copper and contamination of the ores with altered ores from the fire zone of the mine.

In view of the fine-grinding problems involved in the treatment of all of the ores their resistance to crushing is of great interest. Means have not been available for obtaining other than relative data covering this factor. Such data were obtained from laboratory grinding tests and pilot-mill runs before the concentrator was constructed. Crushing of the various ores through 3-mesh size had been practiced for some time in connection with smelting operations, and this work indicated that both the schist and massive sulphides were quite resistant to crushing. The schist ores, though not hard, were slabby, and the massives fine grained and tough. No attempt was made to develop an index as to crushing resistance in this step, inasmudh as economical crushing had been demonstrated. The laboratory tests cited above were concerned with the accumulati of comparative fine-grinding data on representative local ores and ores from several of the operating mines of the Southwest. Briefly, this work demonstrated that United Verde schists and massives were more easily reduced through a given mesh size than the other ores tested--the massives being the softer. The following table lists the ores tested in the order of increasing resistance to crushing, as determined by this test work.

1. United Verde massive
2. United Verde schist
3. Copper Queen
4. Inspiration
5. Nevada Consolidated (McGill)
6. Miami
7. Moctezuma
8. New Cornelia

In so far as the United Verde ores are concerned, it is of interest that plant-scale tests recently conducted to secure similar data have confirmed laboratory results.

Because of the degree of dissemination of the valuable minerals in the ores it has been found expedient to do considerable microscopic work to guide metallurgical practice. This work has been confined chiefly to the study of scre sizes of mill products, using methods developed by the Bureau of Mines station at Salt Lake City, and modified to serve local needs, and has been carried out by the research department of this company. It has had as its object determination of the degree of locking of the valuable minerals with each other and with the gangue and pyrite. In as much as the mineralogy of the ores is comparatively simple screen analyses of products permit calculation of mineralogy, and microscopic study of the sizes permits an estimate of the importance of locking. In effect, detailed quantitative microscopic studies are unnecessary. In reporting on flotation feed and samples of tailings the microscopist directs his attention to locking of chalcopyrite, reporting the percentage locked and the locking mineral. In the examination of concentrates condition of the pyrite is of importance but in this case the degree of freedom is reported. It is important to point out that such estimates of locking or freedom involve certain probability laws and the determination of the extent to which observed conditions must be discounted to arrive at a true picture of locking conditions gives rise to many questions. It is sufficient to say that all free grains observed are not free, and the nature of the locking may determine the chances of a locked grain being shown as free when briquetted. When it is also considered that flotation phenomena have to do with the surface conditions of the mineral particles the necessity for caution in interpretation of the results from microscopic studies is emphasized further.

LOCATION, DESIGN, AND CONSTRUCTION

The United Verde concentrator is at Clarkdale, Yavapai County, Ariz.; the point at which all the United Verde smelting operations are conducted. Geographically it is located on the Verde River at an elevation of 3,400 feet above sea level and at a distance of approximately 6 miles, by standard-guage railroad, from the main haulage level of the United Verde mine. The mine is at Jerome, but the haulage level emerges at Hopewell at an elevation of about 4,500 feet. A subsidiary corporation, called the Verde Tunnel & Smelter Railroad, owns and operates the interconnecting railroad.

The location for the concentrator was selected adjacent to the crushing plant. It should be stated that the ground profile is uphill from the crushing plant to the concentrator, which necessitates delivery of the crushed ore from a lower to a higher elevation, which is different from usual practice. In this case, however, there was no other suitable location. It is interesting to note that the increment of extra cost for delivering the ore uphill is not over a quarter of a cent per ton.

The general location of the concentrating plant is such that tailings can be delivered by gravity through a pipe line to a very desirable location for tailings disposal. The plant is also located favorably with respect to all services such as railroad, water, power, and the like. The railroad facilities, as above mentioned, are provided by the Verde Tunnel & Smelter Railroad, which, however, interconnects with the Atcheson Topeka & Santa Fe Railroad at Clarkdale. The water supply has its source in springs, artesian wells, and a tunnel under the Verde River. Power is available from the smelter power plant, the generators therein being operated on waste-heat steam from reverberatory furnaces. The smelter power system, however, is interconnected with the Arizona Power Co.

The type of construction is that usually found in the concentrating plants of the larger companies in the Southwest, being concrete, steel, and galvanized iron with an abundance of windows for lighting. Preliminary testing of the ore was done by the local staff at Clarkdale, although these tests were supplemented by confirmatory laboratory tests by the Timber Butte Mining Co. Instead of pilot-mill operations a batch of several thousand tons of ore was run through a small concentrating plant at Humboldt, Ariz. Mr. W. N. Rossberg, manager of the Timber Butte Mining Co., contributed valuable advice and consulting service on the problem. Design of the concentrator was initiated by Mr. H. Kenyon Burch, a prominent mill designer of the Southwest, and was continued locally under the direction of the United Verde's chief engineer, Mr. J. E. Lanning.

Expenditure for the concentrating plant did not include the expense of a crushing department, because this department had already been functioning and was of sufficient capacity to crush the amount of feed required by the concentrating department. In this particular set-up the concentrator has been in the nature of a minor department with respect to the smelter, and the crushing plant was designed in a flexible arrangement to permit preparation of ores and fluxes to meet a variety of conditions in the smelter departments. Although the crushing plant has been very efficient from the smelter point of view, it is not an ideal crushing plant for serving a concentrator; some small excess crushing cost, therefore, has to be borne by the concentrator department, but this is not equivalent to the amount of fixed charges that would result from construction of a crushing plant for the concentrator service alone.

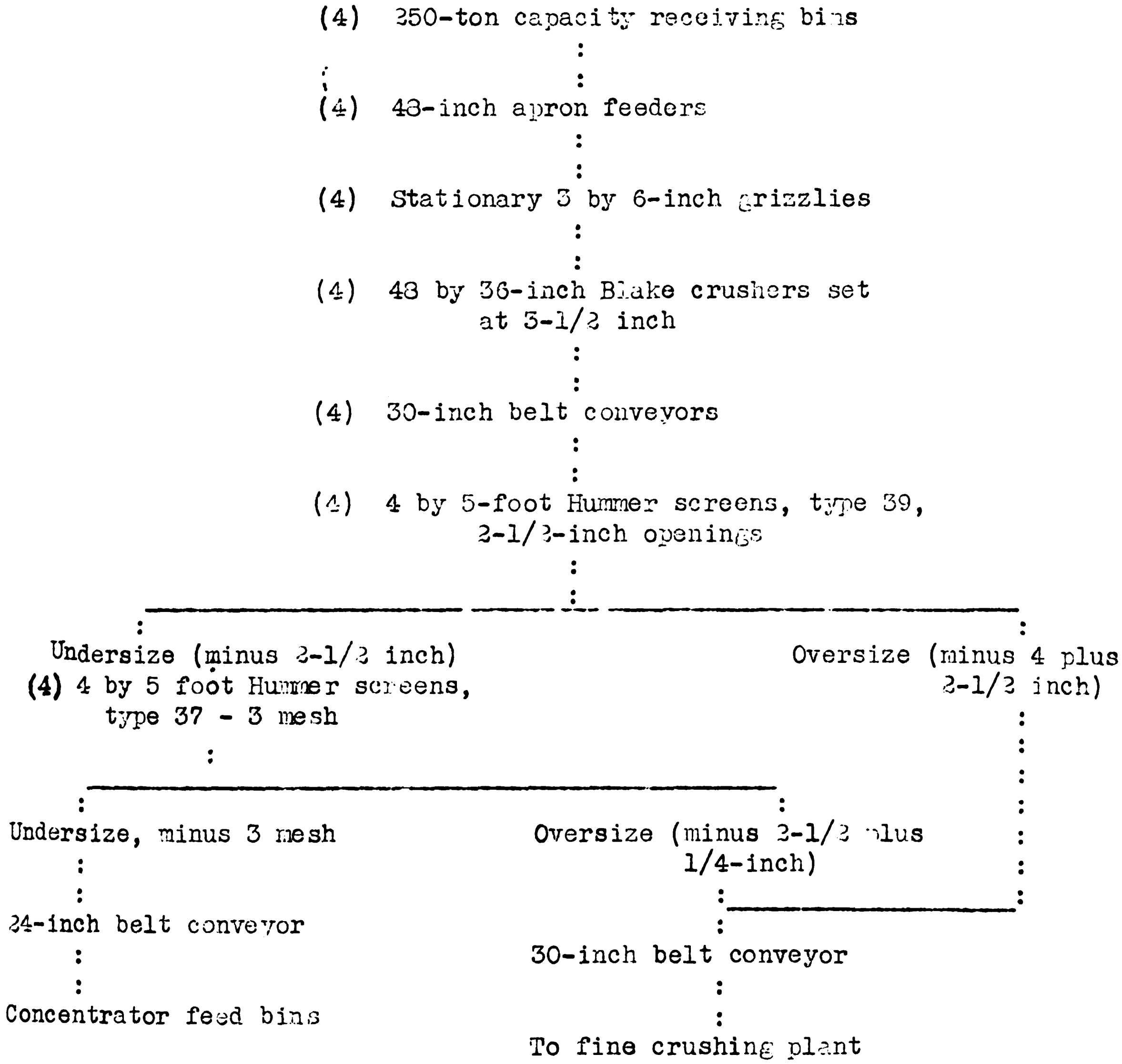
PRESENT METHODS OF CONCENTRATING

Breaking and Intermediate Crushing

The primary breaking is done by Allis-Chalmers jaw crushers, 48 by 36 inches, of all-steel construction. The intermediate crushing is done by 48-inch vertical-type Symons disk crushers, followed by finishing 56 by 24 inch Traylor rolls crushing to minus 1/4-inch size. A system of belt conveyors and elevators is used between units. Figure 1 is the flow sheet of the coarse crushing plant when preparing ore for the concentrator. Figure 2 is the flow sheet of the fine crushing plant.

The maximum size piece of ore received at the crushing plant is about 17 inches. The product of the jaw crushers is minus 4-inch size, and the product of the disk crushers is minus 1-inch size. As stated, the roll product is minus 1/4-inch size. As the flow sheets show the removal of undersize by grizzlies and other screens precede each stage of crushing.

Figure 1.--Flow sheet of coarse crushing plant



(4) 250-ton capacity receiving bins

(4) 48-inch apron feeders

(4) Stationary 3 by 6-inch grizzlies

(4) 48 by 36-inch Blake crushers set
at 3-1/2 inch

(4) 30-inch belt conveyors

(4) 4 by 5-foot Hummer screens, type 39,
2-1/2-inch openings

Undersize (minus 2-1/2 inch)
(4) 4 by 5 foot Hummer screens,
type 37 - 3 mesh

Oversize (minus 4 plus
2-1/2 inch)

Undersize, minus 3 mesh

24-inch belt conveyor

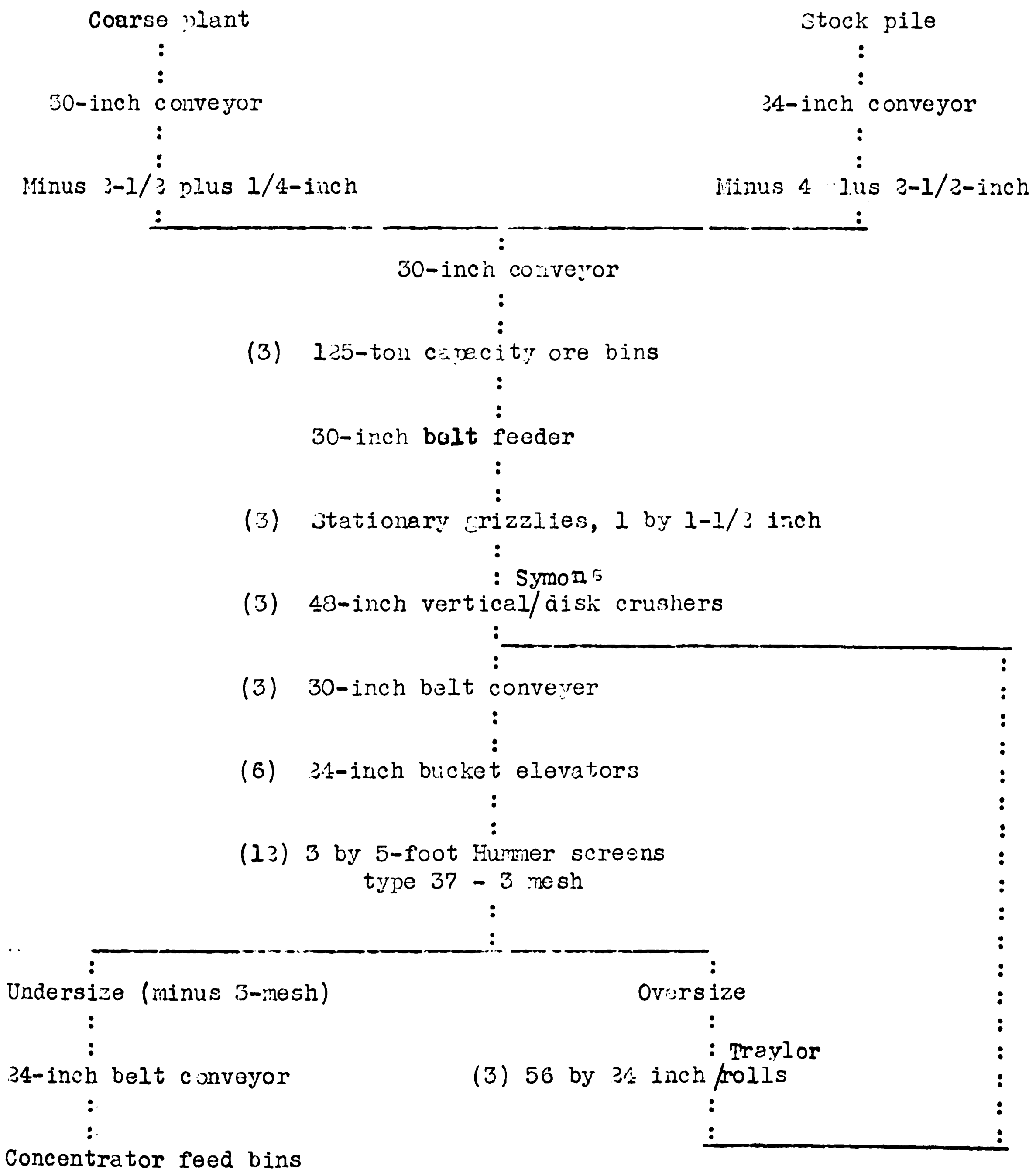
Concentrator feed bins

Oversize (minus 2-1/2 plus
1/4-inch)

30-inch belt conveyor

To fine crushing plant

Figure 2.--Flow sheet of fine-crushing plant


Coarse plant
30-inch conveyor
Minus 2-1/2 plus 1/4-inch
Stock pile
24-inch conveyor
Minus 4 plus 2-1/2-inch
30-inch conveyor
(3) 125-ton capacity ore bins
30-inch belt feeder
(3) Stationary grizzlies, 1 by 1-1/2 inch
(3) 48-inch vertical Symons disk crushers
(3) 30-inch belt conveyer
(6) 24-inch bucket elevators
(12) 3 by 5-foot Hummer screens type 37 - 3 mesh
Undersize (minus 3-mesh)
24-inch belt conveyor
Concentrator feed bins
Oversize
(3) 56 by 24 inch Traylor rolls

CONVEYING

The location of the concentrator with reference to the crushing plant, the source of its feed, necessitates an extensive conveying system for ore transportation. Routing of the ore is as follows: a 36-inch belt conveyor, 506 feet 9 inches long, center to center, traveling 406 feet per minute, receives the ore from the fine crushing plant and delivers it to a second 36-inch conveyor, [illegible]30 feet 7 inches long, center to center, traveling at the rate of 362 feet per minute. The second conveyor is equipped with a motor-driven tripper which delivers the ore to the concentrator bin. The bin comprises two halves , which are filled and emptied alternately. Ore is withdrawn from the bins by drum-type feeders, which deliver the ore to four 24-inch conveyors traveling at the rate of 150 feet per minute. There are 4 such feeders, spaced at 5-foot 6-inch intervals along the bottom of the bin, feeding each of the four conveyors. Two of the conveyors receiving ore from the drum-type feeders delivers to a cross conveyor feeding the primary ball mill of one unit, and the remaining two conveyors receiving ore from the drum feeders deliver to another cross conveyor feeding the other primary mill. These cross conveyors are 24 inches wide and travel at the rate of 36 feet per minute. A Merrick weightometer is installed on the 36-inch conveyor receiving ore from the fine crushing plant, and similar weightometers are also installed on the two 24-inch belts feeding the primary mills. Drum feeders are driven through gang-worm drivers by variable speed motors. The driving arrangement is such that the 4 feeders serving a given ball mill constitute a group in the gang drive. Feeding rate is controlled by the number of feeders operated, finger clutches being available at each feeder, the speed of rotation of the drums and the size of opening at the drum discharge.

Mill flow sheets

As initially constructed the mill consisted of two units, each with a flow sheet as shown in Figure 3. The milling program contemplated an initial milling capacity of 1,000 tons of schist ore per day. Involved in the design, however, were provisions for flexibility in ball-mill speeds which permitted, ultimately, the treatment of 1,200 tons per day (600 tons per unit). Feed sizing was approximately that shown in Table 1,and grinding was carried to 82 per cent minus 100 mesh.

A laboratory investigation of the possibilities for improvement of metallurgy through the rejection of pyrite demonstrated the need for finer grinding--this being particularly necessary as the sulphide content of the feed was raised by admixture of massive sulphides with the schist ores. It was also demonstrated that fine grinding of the total ore, as distinguished

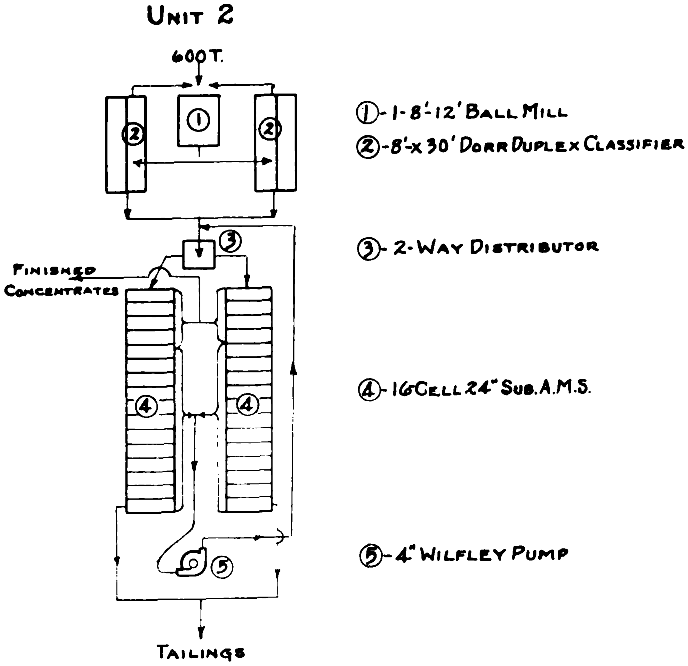


Figure 3. – Single-stage grinding flow sheet.

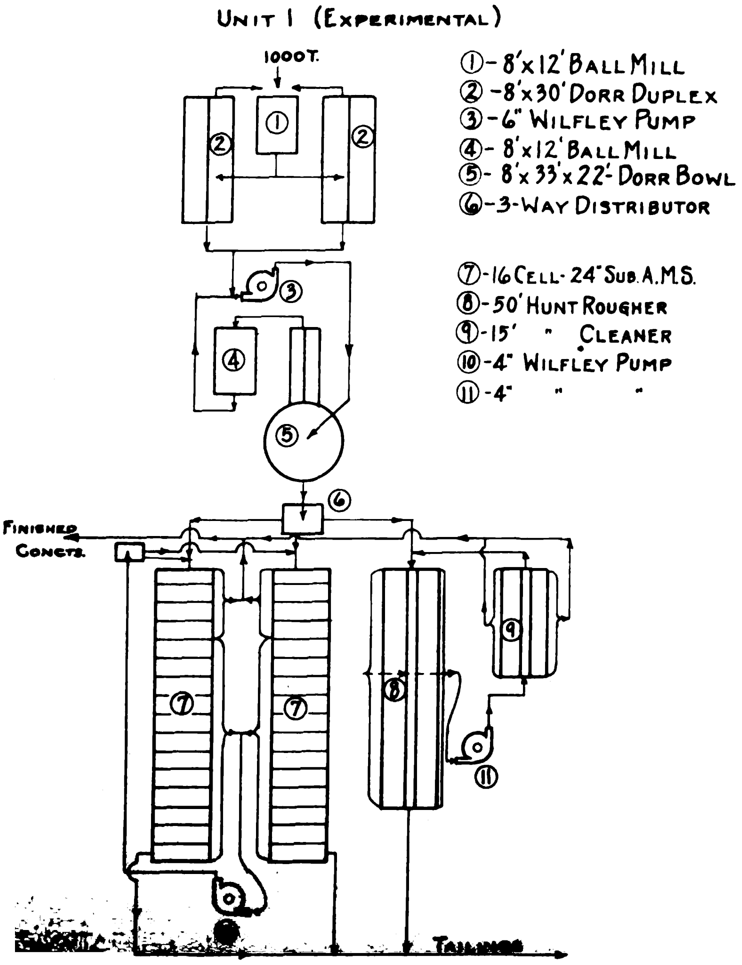


Figure 4. – Two-stage grinding flow sheet

from regrinding of concentrates, was essential, or at least that initial grinding should be carried much farther than is usually the case where regrinding of concentrates is done. Furtherance of this investigation on a mill scale, together with a need for determining crushing resistance of certain of the massive sulphides, necessitated installation of additional grinding equipment, with auxiliaries, on one unit. Although restricted floor space dictated, to a certain extent, placement and units of equipment installed, considerable flexibility of flow sheet was permissible, and means were available for investigation of the several possibilities involved in a finer grinding program. The flow sheet ultimately used on this experimental unit is shown in Figure 4 Conclusions drawn from this test work may be summarized briefly as follows:

(1) Two-stage grinding was more efficient than single-stage grinding.
(2) Large circulating loads through the ball mills are requisite to maximum grinding.
(3) Classifier equipment was inadequate to effectively load the ball mills.

Upon conclusion of the above test program it became desirable to utilize efficiently available grinding equipment in the current milling program. Figure 5 shows the flow sheet adopted to accomplish this.

It will be noted that in this flow sheet new feed goes directly to the ball-mill scoop boxes. Removal of initial fines in the new ore, though offering possibilities, has not, as yet, been tried. With the flow sheet as shown in Figure 5 mill capacity varies from 1,300 to 1,600 tons per day, depending upon the degree of grinding required, which is in turn determined by the character of the ore and the metallurgical objectives sought. Use of the grinding equipment in such a flow sheet provides for maximum utilization of the primary ball mills, although, due to inadequate raking capacity of the bowl classifier, the secondary ball mill is underloaded. Notwithstanding, the following comparison demonstrates the merits of the 2-stage grinding flow.

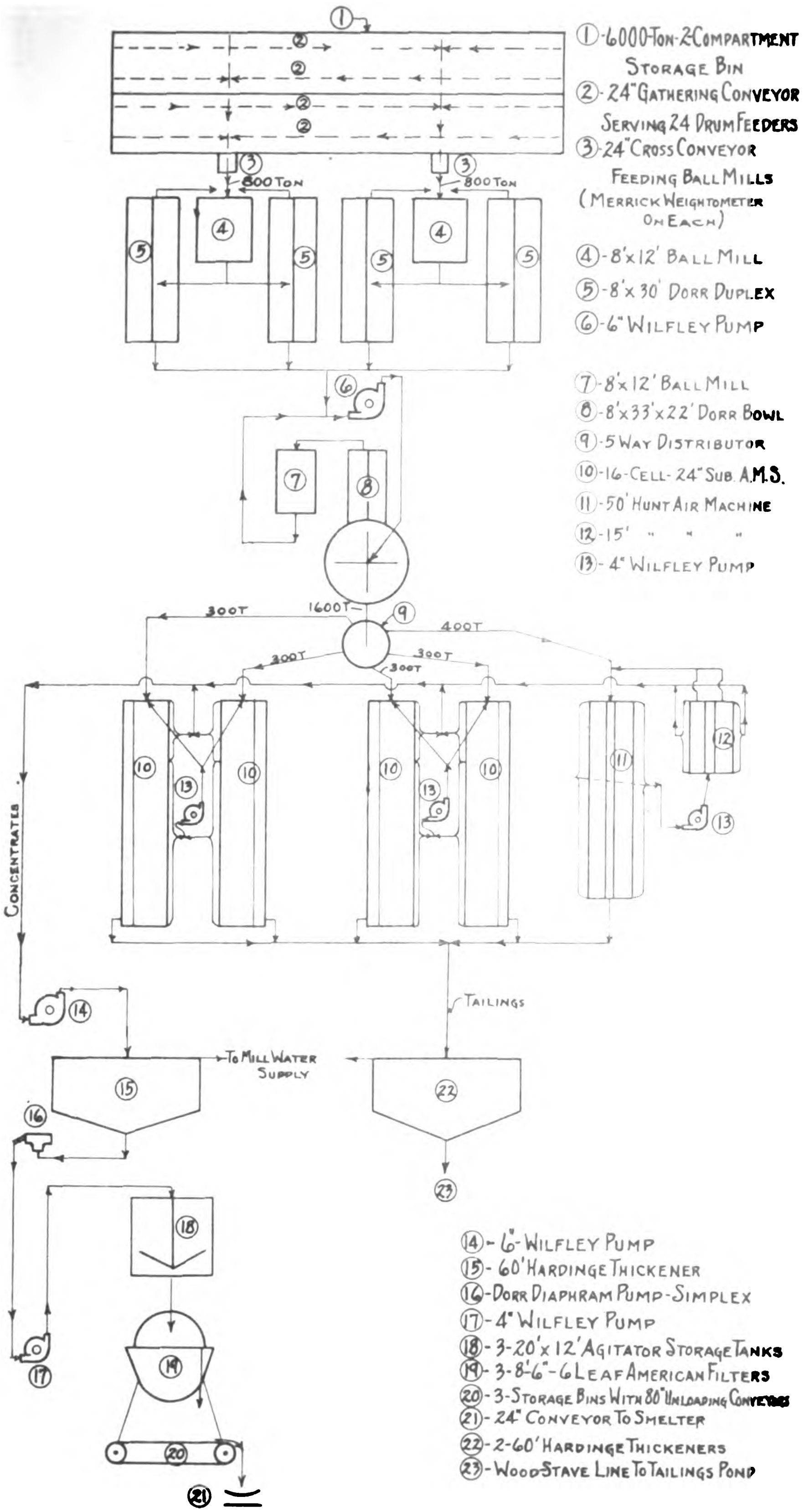

① - 6000-Ton-2-Compartment Storage Bin
② - 24" Gathering Conveyor Serving 24 Drum Feeders
③ - 24" Cross Conveyor Feeding Ball Mills (Merrick Weightometer On Each)
④ - 8'x12' Ball Mill
⑤ - 8'x30' Dorr Duplex
⑥ - 6" Wilfley Pump
⑦ - 8'x12' Ball Mill
⑧ - 8'x33'x22' Dorr Bowl
⑨ - 5 Way Distributor
⑩ - 16-Cell-24" Sub. A.M.S.
⑪ - 50' Hunt Air Machine
⑫ - 15' " "
⑬ - 4" Wilfley Pump
⑭ - 6" Wilfley Pump
⑮ - 60' Hardinge Thickener
⑯ - Dorr Diaphram Pump-Simplex
⑰ - 4" Wilfley Pump
⑱ - 3-20'x12' Agitator Storage Tanks
⑲ - 3-8'-6"-6 Leaf American Filters
⑳ - 3-Storage Bins With 80" Unloading Conveyors
㉑ - 24" Conveyor To Smelter
㉒ - 2-60' Hardinge Thickeners
㉓ - Wood Stave Line To Tailings Pond
800 Ton
800 Ton
1600 T
300 T
400 T
300 T
300 T
300 T
Concentrates
Tailings
To Mill Water Supply

Figure 5. – Mill flow sheet

	Tonnage rate per 24 hours	Tons -100mesh product produced	Kw.h. per ton of -100 product produced
Single Stage	1,200	840	11.4
Two stage	1,600	1,280	10.6

The average tonnage rate maintained during September was 1,532 dry tons per day, with a maximum of 1,613 and a minimum of 1,165 tons.

Fine grinding

Figure 5 is a flow sheet of both primary and secondary grinding stages. All ball mills are approximately 7 feet 6 inches by 11 feet 6 inches inside dimensions, and all are driven by synchronous motors. The primary motors are 400-hp. internal magnetic clutch type and the secondary, 350-hp. direct connected. The selection of synchronous motors for this service was dictated in part by their value for power-factor correction in the local power system. Primary mills are driven at 18 R.P.M. and the secondary at 16 R.P.M. It is necessary to point out that these speeds are not accepted as the most efficient for mills of this size but have proved better than lower speeds. An investigation of this phase of the local grinding problem is under way. Ball-mill gears and pinions are machined steel-spur type. Balls and liners are of white iron and are cast in the local foundry. The ball charge carried in both types of mills approximates 33 tons. Wave-type liners are used and average 3 inches in thickness; there are 20 minor longitudinal waves about the shell periphery. Total weight of liners approximates 25 tons. Make-up ball charges to primaries are approximately 3-1/4 inches in diameter and to the secondary 2-1/4 inches. Under present grinding conditions ball consumption averages 2.0 pounds and liner consumption 0.25 pounds per ton of ore milled. Table 3 gives data as to operating time and tonnage rates and Table 1 the screen sizes of various grinding circuit products together with tonnages and densities on the basis of 1,600 tons daily capacity.

Classification

Dorr classifiers are used in both grinding stages, standard duplex machines being used in closed circuit with the primary mills and a bowl classifier taking the overflows of the primary mill classifiers and operated in close circuit with the secondary mill. Classifier dimensions are shown on the various flow sheets. Primary classifiers have a slope of 3-1/4 inches per foot and the bowl classifier 2-1/8 inches per foot. Rakes of primaries of unit 1 operate at 29 strokes per minute, those of unit 2 at 26 strokes per minute and the speed of the bowl sweeps and rakes is variable, dependent upon duty. The tonnage data given in Table 1 are derived by calculation and of course depend upon the reliability of sampling and sizing tests and also upon the limiting screen size used as a basis for the calculation. In the main, errors are probably not greater than 10 per cent, although differences in the tonnage of sand on units 1 and 2 indicate a wider discrepancy. Difference in raking speeds of the two sets of classifiers will account for some of the difference and operating conditions, other than that for the remainder. The table shows further that although circulation through the primary mills is high, it is low through the secondary, a lack of classifiers capacity being indicated. Classifier sand launders have a slope of 4 inches per foot, but feed launders are flatter, sloping approximately 2 inches per foot. Both launder systems are lined with rubber to minimize wear and improve pulp flow.

Flotation

The flow sheets used with the several machines are shown in Figure 5. That applied on the Minerals Separation machines, consisting of the introduction of feed into the first cell, the return of middlings froth to that cell, and the removal of finished concentrates from the first four cells has proved most suitable both from the viewpoints of metallurgy and operation This is due in part to the fact that unusual frothing conditions have been encountered with the use of the lime circuit. The unusual character of the froth is attributed to contamination of the pulp by reaction products derived from the action of caustic pulp water on the wood-pulp content of the ores. The aluminous nature of the gangue and the presence of aluminum sulphate in the ore are contributing factors. Removal of the wood pulp from the ore prior to its introduction into the grinding circuit is essential and presents a problem for which a solution has not as yet been found.

Four 16-cell 24-inch subaeration Minerals Separation mechanical flotation machines constituted the initial flotation equipment of the concentrator. The Hunt-type air machine was installed for experimental purposes. The flotation machines were chosen because of their recognized utility in the treatment of coarse feed and ores of high specific gravity. At the time of installation, too, the importance of mechanical agitation in the flotation process could not be ignored. Each machine is driven through a Lennix-type drive by a 75 hp. motor. Air is supplied to the four machines at 3 pounds pressure by a blower with a rated capacity of 3,200 cubic feet of free air per minute. Since the addition of the experimental air machine and with the treatment of increased tonnages of ore, air requirements for the total plant approximate 7,000 cubic feet per minute. This is reflected in the distribution of costs and power as shown in Tables 4 and 5.

Reagents

Lime, cyanide, Na-ethyl xanthate, K-amyl xanthate, and steam-distilled pine oil are the reagents in general use. A brief discussion of methods of addition and of the functions attributed to the several reagents follows:

Lump-burned lime of high grade is available in the vicinity and is purchased on an available CaO basis. It is slacked in the plant and fed as milk of lime to the primary ball mills. The feeding device, developed locally, is shown in Figure 6. The lime neutralizes deleterious soluble ore salts and is effective for pyrite depression. Its addition to the primary ball mills is essential for best copper recovery and pyrite rejection. It is used in sufficient quantities to provide between 0.5 and 1.0 pound of excess CaO per ton of water in the flotation feed. Consumption ranges between 5 and 20 pounds per ton of ore, dependent largely upon soluble salt content of the ores. Titrations of classifier overflow and flotation feed waters with standard acid solutions, using phenolpthalein as an indicator, are the basis of causticity control. Close control is not possible, owing to the variable nature of the ores treated, and this in the main dictates the use of higher causticities than are necessary to obtain requisite rejection of pyrite in the flotation operations.

Cyanide is added to the ball-mill feed as crude calcium cyanide by a feeder provided by the cyanide manufacturer. It is used in minor amounts only and appears to function as an activator of chalcopyrite, though effective to a small degree for pyrite depression.

The two xanthates are used in variable amounts, being fed by chain-bucket feeders to the bowl-classifier overflow. They are fed in controlled proportions, as demanded by frothing conditions. The amyl xanthate has proven the better collector and causes marked flocculation of the floatable sulphides. This quality is utilized in the control of frothing although its sole use is seldom permissible because of resulting over flocculation of sulphides and consequent destruction of froth.

Steam-distilled pine oil is the frother and is fed by means of drip buckets to various points in the flotation circuit. Other feeding devices have been and are being experimented with; but points of addition are widely separated, and quantities of pine oil added are so small that the usual feeders have proved inadequate. The feeding method offering most promise comprises use of a pulsating pump supplying a pipe line equipped with spring-controlled needle valves of the type common to lubrication systems.

The quantities of the several reagents used per ton of ore treated are given in Table 5.

Screen analyses of flotation feed, concentrates, and tailings are given in Table 1a.

Dewatering and handing concentrates

Flotation concentrates of a density ranging between 35 and 40 per cent of solids flow by gravity to a sump serving two 6-inch Wilfley pumps. One of these pumps is operated continuously, the other serving as a stand by, and delivers the concentrates to a 60 by 8-foot 3-inch Hardinge thickener The thickener overflow is diverted to the reclaimed water system and thence to the mill reservoir. Thickened concentrates are withdrawn from the thickener, at a density of 70 to 75 per cent of solids, by a Dorrco diaphragm pump and delivered through launders to a pump sump serving two 4-inch Wilfley pumps. Pumping service here is intermittent, and one pump suffices for the delivery of the concentrates to agitator--equipped storage tanks above the filters. The storage of thickened concentrates is necessary to permit filtering and delivery to the smelting department as needed. The storage tanks deliver to three American-type filters of 6-leaf, 10-sector design, 8 feet 6 inches in diameter. Each filter provides 600 square feet of filtering area. All filters are operated simultaneously to expedite filtering of concentrates and disposal of cake. The total filtering time is thus limited to approximately 6 hours per day. Filter capacity averages 0.75 ton of concentrates per square foot of filtering surface per 24 hours. Vacuum is maintained at 22.5 inches by Ingersoll-Rand Imperial type XB belt-driven vacuum pumps. Cylinder dimensions of pumps are 27 by 14 inches.

Pumps are driven at 170 R.P.M. by 100-horsepower motors. Blow air for the filters is obtained from the flotation blower system at 1.5 pounds pressure. Filter bags are of the standard filter twill, single ply, and are purchased from the filter manufacturer. The life of bags, measured in total elapsed time of service, is 83 days, which in terms of continuous service approximates 20 days. The moisture content of the filter cake ranges between 12 and 14 per cent and appears to depend upon the alumina content of the concentrates, which ranges from 2.3 to 3.5 per cent. Filtrate water from the barometric leg is automatically reclaimed and returned to the concentrates,thickener by 2-inch centrifugal pumps, float-controlled from the barometric leg sump.

Each filter discharges its cake to a storage bin surmounting two unloading conveyors of special design, combining elements of both the pan and belt type of conveyors. Concentrates storage capacity available in the three bins approximates 150 tons. The unloading conveyors discharge through chutes to a 24-inch belt conveyor, which in turn delivers to the roaster-charging conveyor system of the smelter. The conveyor system is equipped with a Merrick weightometer for recording the tonnage of concentrates handled. To increase accuracy of weighing a uniform ribbon of concentrates is maintained on the belt by the use of wire cake cutters at the discharge point of the unloading conveyors feeding it.

Dewatering and disposal of tailings

Tailings pulp flows by gravity to a distributor adjacent to the thickener installation, where its distribution to the thickeners and to the wood-stave disposal line is accomplished. Two 60-foot by 8-foot 3-inch Hardinge thickeners are available for thickening tailings, but their capacity is limited to something under 50 per cent of the tailings tonnage made at the milling rate prevailing during the period covered by this report. Tailings density averages 25 per cent of solids as rejected by the flotation machines, and this constitutes thickener feed. Thickeners discharge by gravity through bushing-controlled spigots at the intake of the wood-stave line. Discharge density approximates 50 to 55 per cent of solids and feed to the tailings line 35 to 40 per cent of solids. Thickeners are operated to a clear overflow, which is diverted to the reclaimed water system. Delivery of the tailings to the tailings pond is accomplished through a wood-stave gravity line, which has been in service over two and one-half years and shows but little wear--, as practically no repairs having been required during that period.

The tailings pond site is an ancient channel and valley of the Verde River and offers an ideal storage area. The terrain provides natural closure of three sides of the area. A trestle was erected across this channel and carries a launder which provides for disposal of tailings and the building of a dam beneath it thus effecting closure of the fourth side of the area. Disposal of tailings on the dam is accomplished through 1.5 inch holes drilled in the bottom of the launder. Deposition on the dam is restricted to day shift only and then to definite areas, the launder spigots not in use being closed by wooden plugs. Tailings not diverted to dam building are by-passed to the sides and center of the pond area through ditches and launders on either side of the channel or valley. The dam face is raised by hand shoveling, a minor border or bank being thrown up to prevent back-flow of tailings over the dam face during the period of spigot discharge from the launder above. By this operation elevation of the dam can be increased more rapidly than the general level of the pond, and no difficulties have been encountered in keeping the tailings impounded. Under present conditions the entire area can be readily wetted by proper disposition of talings, but this had not been necessary because the protected position of the pond together with a tendency of the tailings to crust or cake on drying have sufficed to eliminate the dust nuisance.

Sampling

With minor exceptions it has not been difficult to obtain accurate samples. Automcatic samplers actuated through cables by a master water-tilting box were part of the initial equipment of the concentrator. Maintenance of this mechanism in proper working order, particularly as regards regulation of the travel of the several samplers driven by it, proved difficult, and adoption of hand sampling was necessitated. This practice has been continued up to the present time with one exception--an automatic sampler is operated on the general tailings launder. This machine samples all mill rejects, and the sample thus obtained is used for determination of mill losses. It also serves as a check on the hand sampling of tailings on the mill units.

All control samples are cut from pulp streams, except, of course, the moisture sample of mill feed and that of the filter cake. Pulp streams are in general small, and with adequate sampling steps grab samples have proved reliable under the prevailing fine-grinding conditions. The sampling interval is 15 minutes, and samples are taken by samplers assigned to each operating shift by the metallurgical department.

Concentrator samples are prepared for analysis at the concentrator bucking room, but analyses are made in the general laboratory of the smelter. Bucking-room duties involve the determination of wet and dry weights of samples and the other necessary steps in the preparation of pulps for analysis. The bulk of the water is removed from samples of pulp in pressure filters. The filter cake is then dried for a period of approximately 1.5 hours in electric ovens, whose temperature is thermostatically controlled between 340 and 360° F

Riffling of samples to desired size, screening, and bucking follow, after which the samples are placed in appropriately marked 8-ounce paper bags for delivery to the laboratory. Preparation of monthly composites and the making of sizing tests are other duties of the sampling force.

Water supply

The concentrator water-supply system is an integral part of the local smelter water system. Adjacent to the concentrator and supplying it with water is a reservoir of approximately 600,000 gallons capacity, which is supplied from the smelter water supply system. The latter system derives its water from two sources, springs in the adjacent mountains and the Verde River. Spring water flows to the system by gravity. The river water must be pumped. Pumping equipment available for concentrater service consists of the following: Two centrifugal pumps, each with a capacity of 600 gallons per minute and two triplex plunger pumps with a capacity of 400 gallons per minute. Pumps operate against a static head of 290 feet.

Water reclaimed in concentrator thickening operations is also returned to the reservoir. Thickener overflows are collected in two tanks of 12,000 gallons capacity each, which serve as sumps for two centrifugal pumps of 1,000 gallons per minute capacity direct connected to 50 hp., 1,750 R.P.M. motors. Pumps are automatically operated by float switches. But one pump is operated at a time, the other serving as a stand-by. Pumps operate against a static head of 120 feet.

Available water may be properly classed as "hard", but does not contain elements that interfere with ore-dressing practices. Mixture of fresh and reclaimed waters has been found undesirable because of precipitation resulting from reactions between excess causticity of the reclaimed waters and hardness of the fresh water. The precipitate accumulates in large quantity in the reservoir, and precipitation is extended into mill-supply lines, with the result that scale and accretions form in undesirable amounts in those lines. Separation of the two waters up to the point of use in the mill is contemplated.

Water consumption data are given in Table 3.

Concentrator recovery

Analyses of concentrator products are given in Table 2 and metallurgical data in Table 3. Significant features of these data are the pyrite content of all of the products and the copper content of the concentrates and tailings. The concentrates are low grade, due to dilution by gangue, pyrite, and marmatite. The screen analysis of these concentrates given in Table 1a presents another interesting feature, namely, decreasing grade with decreasing size of material. Complete analyses of screen sizes from concentrates of other periods have shown that chalcopyrite and gangue are the major constituents of the coarsest size, with pyrite and marmatite becoming of increasing importance with decrease in size of material. Closed-circuit grinding effects are thus reflected in the flotation concentrates. The feature of particular interest, however, is the apparent increased flotation of pyrite in the finer sizes from a circuit in which reagent conditions are such that the rejection of approximately 70 per cent of the pyrite in the feed is accomplished. On the basis of percentage rejection of pyrite from the various sizes of the flotation feed itis not surprising to find, under the screen-analysis conditions shown, that rejection is high in the coarser sizes and low in the finer. From a tonnage standpoint, greatest rejection of pyrite, of course, comes from the finest size, which constitutes the bulk of the flotation feed. Microscopic study of the concentrate sizes, above and including the plus 300-mesh size, resulted in the conclusion that 75 per cent of the pyrite, and also of the marmatite, was free and rejectable. Peculiarly, all attempts made to grade up the concentrate by flotation and manipulation resulted in increased losses of tailings, which were out of proportion to the rejection of pyrite accomplished, and such grading-up could not be achieved without grinding of the concentrates prior to further flotation treatment.

Turning to the tailings screen analysis, distribution of copper values indicates need for finer grinding if recoveries are to be improved. This conclusion is verified by microscopic studies of the sizes, above and including the plus 300-mesh size, from which the conclusion was drawn that chalcopyrite present was largely locked either to gangue or to pyrite. Considering this condition in conjunction with the studies of concentrates it would appear that either a certain type of pyrite present in the ore exhibits the same flotability as the chalcopyrite (or possibly certain chalcopyrite-pyrite middlings), or the entire problem is a middlings problem and conclusions drawn from microscopic studies of the concentrates are erroneous. In view of the results of regrinding tests on such concentrates the latter conclusion would appear to be the correct one.

The zinc content of the concentrates has directed attention to the possibilities for improving grade of concentrates through rejection and separation of the marmatite. Microscopic studies of polished sections of the ores indicated that association of chalcopyrite and marmatite was more intimate than that between the chalcopyrite and the pyrite. Coupled with that is the fact that marmatite in the current ore is highly activated and as yet, no means have been developed for depressing it.

It is important to point out that the considerations involved in the preceding discussion bear directly on the local grinding problems. They serve to emphasize the importance of a fine initial grind on the United Verde ores for, although a regrind practice for concentrates may prove of economic value, the outstanding consideration appears to be that initial grinding must be carried to such a point that middlings if remaining must be flotable in initial roughing flotation operations. It also follows that as the sulphide content of the ores increases, assuming similar conditions of dissemination of the valuable minerals, the more important will the initial grinding work become.

Summary of costs

Tables 4 and 5 present summaries of milling costs in terms of labor, power, and money units for September, 1929. The total is somewhat higher than average, and individual items may vary somewhat from average figures. Under fine-grinding supplies, balls and liners have been charged out at average consumptinn rates.

Labor costs include a bonus payment to employees for increased efficiency of the group. The bonus system has as its object increase through the medium of reduced treatment costs and increased recovery of values of the net return obtained per ton of ore treated. To employees it offers, as an incentive for higher efficiency in group effort, participation in the yield from such increased efficiency. Payments to employees are based upon predetermined standards of efficiency, covering the items ratio of concentration, recovery of values, and operating time in unit hours, and number of employees. Ratics of concentration and recovery standards are properly balanced to measure net increased in metallurgical efficiency. Operating-time standards make reasonable allowances for repairs and necessitate efficient mechanical maintenance if bonus payments are obtained.

Table 1. - Screen sizes of grinding-circuit samples taken on September 5, 1929, tonnage.
Distribution and consistency of pulps. Ore treated, 1,600 tons.

Mesh	Feed				Ball-mill discharge				Primary-classifier sand			
	Unit 1		Unit 2		Unit 1		Unit 2		Unit 1		Unit 2	
	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent
Plus 3...	3.15	3.15	3.00	3.00	0.1	0.1	0.5	0.5	0.3	0.3	0.3	0.3
Plus 4...	14.75	17.90	15.20	18.20	.6	.7	1.8	2.3	.7	1.0	1.8	2.1
Plus 6...	13.35	31.25	14.90	33.10	.9	1.6	1.5	3.8	1.2	2.2	2.0	4.1
Plus 8...	11.35	42.60	11.70	44.80	1.4	3.0	2.0	5.8	1.9	4.1	2.9	7.0
Plus 10..	10.25	52.85	10.00	54.80	2.2	5.2	2.6	8.4	3.2	7.3	3.8	10.8
Plus 14..	8.45	61.30	8.20	63.00	3.8	9.0	3.7	12.1	4.8	12.1	5.2	16.0
Plus 20..	6.10	67.40	5.80	68.80	5.3	14.3	4.7	16.8	7.0	19.1	6.7	22.7
Plus 28..	5.65	73.05	5.45	74.25	9.5	23.8	8.0	24.8	11.7	30.8	10.8	33.5
Plus 35..	3.75	76.80	3.55	77.80	12.7	36.5	11.7	36.5	15.5	46.3	14.6	48.1
Plus 48..	2.70	79.50	2.70	80.50	13.6	50.1	12.8	49.3	13.4	59.7	14.5	62.6
Plus 65..	2.55	82.05	2.45	82.95	13.4	63.5	11.8	61.1	15.0	74.7	13.1	75.7
Plus 100.	2.55	84.60	2.25	85.20	9.7	73.2	9.7	70.8	9.9	84.6	8.9	84.6
Plus 150.	2.20	86.80	2.10	87.30	6.9	80.1	6.7	77.5	5.2	89.8	5.5	90.1
Plus 200.	1.50	88.30	1.40	88.70	3.2	83.3	3.8	81.3	2.2	92.0	2.0	92.1
Minus 200	11.70	100.00	11.30	100.00	16.7	100.0	18.7	100.0	8.0	100.0	7.9	100.0

Mesh	Primary classifier overflow				Bowl classifier						Secondary ball-mill discharge	
	Unit 1		Unit 2		Feed		Sand		Overflow			
	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent	Per cent	Cum. per cent
Plus 28..	0.1	0.1	0.2	0.2	0.2	0.2	0.7	0.7			0.8	0.8
Plus 35..	1.6	1.7	1.9	2.1	2.0	2.2	4.5	5.2			2.8	3.6
Plus 48..	4.0	5.7	4.4	6.5	4.2	6.4	11.8	17.0			3.6	7.2
Plus 65..	8.0	13.7	8.2	14.7	8.2	14.6	22.0	39.0	0.8	0.8	8.6	15.8
Plus 100.	12.6	26.3	12.6	27.3	16.2	30.8	29.6	68.6	5.6	6.4	22.7	38.5
Plus 150.	9.8	36.1	9.7	37.0	13.5	44.3	16.9	85.5	10.0	16.4	18.5	57.0
Plus 200.	7.3	43.4	7.0	44.0	9.9	54.2	9.4	94.9	9.8	26.2	12.9	69.9
Minus 200	56.6	100.0	56.0	100.0	45.8	100.0	5.1	100.0	73.8	100.0	30.1	100.0

Product	Tons	Per cent solids
Feed, unit 1	800	97.8
Feed, unit 2	800	98.0
Ball-mill discharge, unit 1	4,472	81.8
Ball-mill discharge, unit 2	3,560	80.8
Primary classifier sands, unit 1	3,672	84.3
Primary classifier sands, unit 2	2,760	83.3
Primary classifier overflow, unit 1	800	38.4
Primary classifier overflow, unit 2	800	43.3
Bowl-classifier feed	2,701	41.0
Bowl-classifier sands	1,101	77.2
Bowl-classifier overflow	1,600	25.6
Secondary ball-mill discharge	1,101	77.5

Table 2. - Average mill assays and recoveries for September, 1929.

Assays

Product	Weight, per cent	Per cent										Ounces per ton		Per cent
		Total copper	Sulphide copper	Oxide copper	Acid soluble copper	SiO_2	Al_2O_3	Fe	S	Zn	CaO	Au	Ag	Total
Flotation feed	100.000	3.06	2.89	0.034	0.170	31.2	11.4	20.4	16.1	1.2	1.6	0.0157	0.7379	85.0
Concentrates....	19.591	14.42	14.362	.003	.058	5.2	2.8	30.9	36.2	5.1	.7	.0533	2.6696	95.3
Tailings............	80.409	.323	.252	.036	.071	37.6	13.6	17.9	10.8	.29	1.8	.0070	.2718	82.3

Mineralogical calculations, per cent

Product	$CuFeS_2$	ZnS.FeS[1]	FeS_2	Schist	Quartz	Hydrated iron oxides	$CaCO_3$	Totals
Flotation feed.....................	8.81	1.99	23.17	45.60	18.89	1.55	2.85	102.86
Flotation concentrates..........	41.52	8.46	35.47	11.20	2.18	.28	1.25	100.08
Flotation tailings................	.93	.48	19.32	54.40	22.91	2.47	3.21	103.72

1 - United Verde ZnS.FeS contains 60.3 per cent Zn, 33.3 per cent S, and 6.44 per cent Fe.

Recovery or less in products per cent

Product	Total copper	Sulphide copper	Oxide copper	Acid soluble copper	Zn	Fe	Al_2O_3	S	SiO_2	Ag	Au
Flotation feed	100.000	100.000	100.00	100.00	100.000	100.000	100.00	100.000	100.000	100.00	100.00
Concentrates	91.129	93.919	14.86	66.42	78.915	29.674	4.78	45.752	3.265	70.52	64.98
Tailings............................	8.871	6.081	85.14	33.58	21.085	70.336	95.22	54.248	96.735	29.48	35.02

Table 1a.--Screen sizes and copper analyses of concentrator feed and products

September, 1929, monthly composites
Ore treated, 44,440 tons
Concentrates, produced, 8,706 tons

Mesh	Mill feed		Flotation feed		Flotation concentrates		Flotation tailings	
	Per cent		Per cent		Per cent		Per cent	
	Material	Copper	Material	Copper	Material	Copper	Material	Copper
Plus 3..	8.525	2.38						
Plus 4..	25.325	3.18						
Plus 6..	15.075	3.11						
Plus 8..	11.650	3.03						
Plus 10.	8.350	3.03						
Plus 14.	5.975	3.11						
Plus 20.	4.100	2.99						
Plus 28.	3.475	3.03						
Plus 35.	2.600	3.13						
Plus 48.	1.950	3.16	0.3					
Plus 65.	1.150	3.21	1.1	0.61			1.7	0.330
Plus 100	2.200	3.28	7.2	0.92	1.2		8.5	0.440
Plus 150	1.450	3.54	10.5	1.99	3.5	18.12	12.0	0.490
Plus 200	1.175	3.79	10.6	3.10	9.7	15.94	10.3	0.480
Plus 300 [1]	7.000	3.84	13.1	3.82	18.5	14.17	12.4	0.380
Minus 300			57.2	3.51	67.1	14.17	55.1	0.260
Comp....	100.000	3.10	100.0	3.10	100.0	14.48	100.0	0.342

[1] Mill feed plus 300 is minus 200 mesh

Table 3.--Metallurgical data for September, 1929

Total ore treated	wet tons	44,944
Total ore treated	dry tons	44,440
Time operated[1]	days	29
Time operated per day	hours	24
Average ore treated per 24 hours	tons	1,532
Average sections running		1.892
Average ore per 24 hours per section	tons	783
Total concentrates	"	8,706
Average concentrates per 24 hours	"	300
Total concentrates	"	8,706
Recovery of total copper	per cent	91.129
Recovery of sulphide copper	" "	93.919
Recovery of oxide copper[2]	" "	14.860
Recovery of acid soluble copper[3]	" "	66.42
Recovery of zinc	" "	78.915
Recovery of gold	" "	64.980
Recovery of silver	" "	70.520
Rejection of pyrite	" "	68.82
Rejection of schist	" "	95.22
Rejection of silica	" "	96.735
Ratio of concentration		5.1044
Total water used per ton ore treated	gallons	703
Fresh water used per ton ore treated		367
Reclaimed water	per cent, of total	45.87
Ball consumption per ton of ore treated	pounds	2.028
Liner consumption per ton or ore treated	"	.218

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[1] Mill shut down on Labor Day.

[2] Oxide copper determined by SO_2 leach.

[3] Acid-soluble copper determined by leaching with a 5 per cent solution of sulphuric acid.

Table 4.--Summary of costs for September, 1929

Concentrates produced: 8,706 tons   Ore treated; 44,440 tons.

	Total	Per ton
Receiving ores		
Unloading	$ 422.21	$ 0.0095
Coarse crushing	3,335.77	.0751
Fine crushing	5,245.16	.1180
Total	9,003.14	.2026
Conveying and feeding		
Wages	664.39	.0149
Supplies	15.03	.0003
Electric power	162.02	.0036
Repairs		
Wages	150.76	.0035
Supplies	47.75	.0011
Total	1,039.95	.0234
Grinding		
Wages	788.68	0.0178
Supplies	196.20	.0044
Electric power	3,266.78	.0735
Grinding balls	1,858.49	.0419
Liners	444.67	.0100
Repairs		
Wages	372.38	.0084
Supplies	351.26	.0079
Total	7,278.46	.1639
Flotation		
Wages	957.07	0.0216
Supplies	16.36	.0004
Electric power	1,103.36	.0248
Reagents	5,108.08	.1149
32 ounce air	311.36	.0072
Repairs		
Wages	189.42	.0043
Supplies	109.68	.0025
Total	7,795.33	.1757
Thickening of concentrates		
Wages	525.23	0.0118
Supplies	1.65	.0000
Electric power	162.38	.0037
Repairs		
Wages	53.75	.0012
Supplies	43.46	.0010
Total	786.47	.0177

Table 4.--Summary of costs for September 1929. (Continued)

	Total	Per Ton
Filtering		
Wages	$ 365.57	$ 0.0082
Supplies	3.73	.0001
Electric power	125.85	.0028
32 ounce air	10.40	.0003
Repairs		
Wages	52.06	.0012
Supplies	142.62	.0033
Total	703.23	.0159
Handling of concentrates		
Wages	49.25	0.0011
Supplies	.37	.0000
Electric power	53.03	.0012
Repairs		
Wages	90.30	.0020
Supplies	49.99	.0011
Total	242.94	.0054
Tailings disposal		
Wages	158.77	0.0036
Supplies	6.38	.0002
Repairs		
Wages	31.13	.0007
Supplies	3.96	.0001
Total	200.54	.0046
Mill water		
Electric power	51.75	0.0012
Fresh water supply	1,128.40	.0248
Repairs		
Wages	43.41	.0010
Supplies	21.52	.0005
Total	1,245.08	.0275
Routine sampling and testing		
Wages	1,364.09	0.0307
Supplies	39.67	.0009
Electric power	55.76	.0013
Repairs		
Wages	9.57	.0002
Supplies	2.02	.0000
Total	1,471.11	.0331

Table 4.--Summary of costs for September 1929.

(Concluded)

	Total	Per ton
Concentrator unclassified		
Wages	$ 2,076.70	$ 0.0467
Supplies	153.84	.0035
High-pressure air	17.63	.0004
Electric power	148.20	.0033
Mineral separation royalty	444.40	.0100
Repairs		
Wages	64.20	.0014
Supplies	59.88	.0014
Total	2,964.85	.0667
Redistributable accounts		
32 ounce air		
Power	315.11	
Repairs	6.65	
Total	321.76	
Distribution of 32 ounce air		
Flotation	311.36	
Filtering	10.40	
Total	321.76	

Total cost of concentrating 44,440 tons of ore $32,731.10

Total cost per ton .... .7365

Power costs, 0.75 cent per kw. h.

Table 5.--Summary of costs in units of labor, power, and supplies, September 1929.

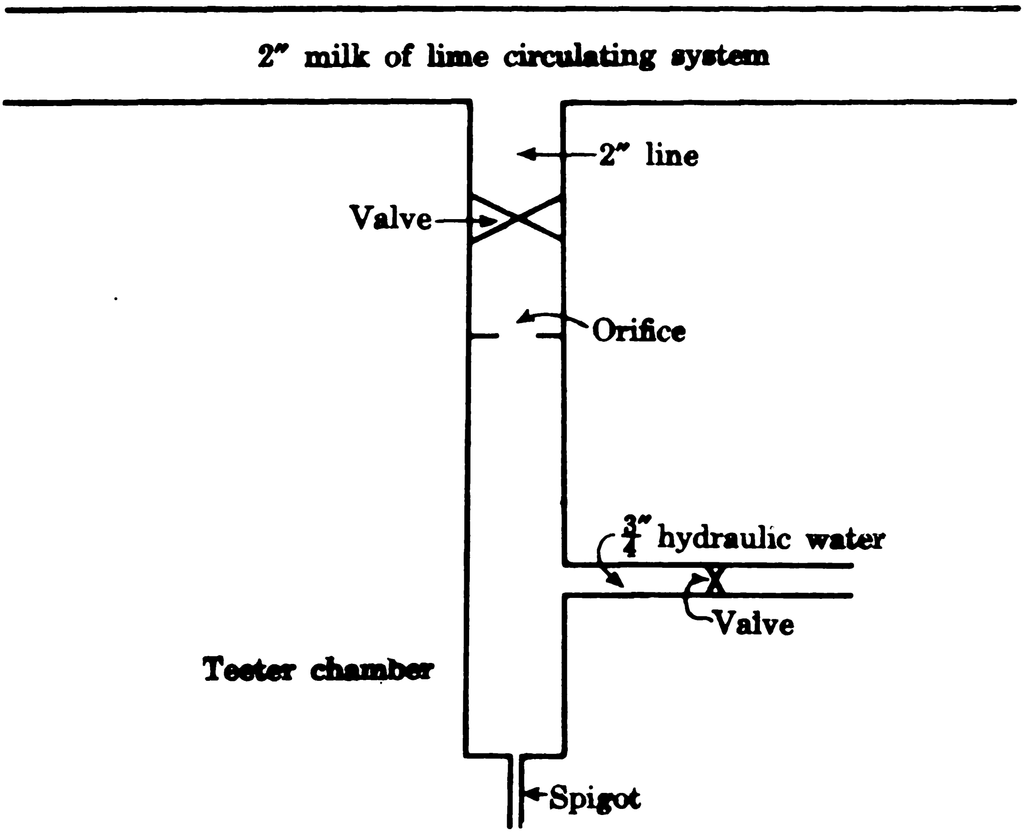
Ore treated: 44,440 tons

Concentrates produced: 8,706 tons

	Operation	Repairs	Total
A. Labor (man-shifts, excluding crushing)			
Conveying and feeding	167.00	29.05	196.05
Grinding	129.00	66.55	195.55
Flotation	155.50	31.89	187.39
Thickening	95.50	9.19	104.69
Filtering	60.50	8.50	69.00
Handling of concentrates	9.00	15.64	24.64
Tailings disposal	29.00	5.24	34.25
Mill water	10.00	7.34	17.34
Sampling and testing	194.00	1.40	195.40
Unclassified	110.00	10.50	120.50
Totals	959.50	185.30	1,144.80
Tons per man shift			38.82

B. Power (kw.h.used)	Total	Per ton	Per cent of total
Conveying and feeding	21,603	0.486	2.976
Grinding	435,571	9.801	60.004
Flotation	147,115	3.310	20.266
Thickening	21,650	.487	2.983
Filtering	16,780	.378	2.312
Handling of concentrates	7,070	.159	.974
Tailings disposal	0	.000	.000
Water	6,900	.155	.951
Sampling and testing	7,435	.167	1.024
Lighting and unclassified	19,760	.445	2.722
32 ounce air	42,015	.946	5.788
Totals	725,900	16.334	100.000

C. Reagents	Pounds	Pounds per ton
Lime, burnt (high calcium)	423,320	9.5371
Cyanide, crude calcium	1,000	.0225
Xanthate, potassium amyl	3,000	.0713
Xanthate, sodium ethyl	3,980	.0896
Pine oil, Yarmor	4,220	.0949
D. Principal supplies		
Grinding balls	90,124	2.028
Liners	9,688	.218
Pump parts	487	.011
Flotation impellors	1,317	.030



NOTE: Hydraulic water supply must be of greater pressure than milk of lime supply and available in sufficient volume to satisfy spigot capacity. Standard pipe fittings are used in assembly

Figure 6. – Milk of lime spigot feeder